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TRANSLATIONS OF ARTICLES ON COMMUNIST CHINA'S NONFERROUS METALS INDUSTRY

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TRANSLATIONS OF ARTICLES ON COMMUNIST CHINA'S NONFERROUS METALS INDUSTRY

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I. SUGGESTIONS REGARDING SAVINGS IN AND RECOVERY OF CHEMICAL TESTING RAEGENTS IN CHEMICAL ANALYSIS AND RESEARCH WORK

/Following is a translation of an unsigned article in Yu-se Chin-shu (Nonferrous Metals), No. 8, Peiping, 18 April 1959, Page 28./

Editorial note: Following the rapid progress in industrial development, chemical analysis work has become increasingly important and, as a consequence, chemical reagent requirements have also risen markedly. How shall we solve the problem? Reliance on the production of testing reagents alone is not enough. We should develop hidden potential, create strength from within, acquire local materials, and find substitutes. We believe that the suggestion of the Laboratory of the No. 1 Lead-Zinc Mine is very good and conforms with the spirit of the main line.

Under the reflection of the Party's main line and the overall leap forward conditions in our country, it is possible that some units are finding temporary difficulties in obtaining enough chemical testing reagents so that chemical analysis and research work in general are affected. To carry out our normal work and overcome the shortage of materials, aside from seeking ways to obtain more supplies, the entire staff and workers of the No. 1 Lead-Zinc Mine Beneficiation Plant Analytical Laboratory, under the leadership of the Party and on the basis of maintaining or improving the quality of chemical analysis work, have implemented the following measures: (1) eliminated the use of nitric acid in testing samples of galena ore with regard to the relative volumes of lead and zinc minerals; (2) reduce the use of nitric acid by 50% in the chemical analyses of the nickel, cobalt, and copper content of nickel ores (sulfide nickeliferous ores); and (3) reduced the consumption of nitric and hydrochloric acids in analyzing the sulfur content of ores. Through these measures, the overall consumption of nitric acid at the laboratory has been reduced by 80% and of hydrochloric acid, by 30%, etc.

These successes raised our confidence in trying to make savings elsewhere. From waste reagent solutions, we recovered sodium chloride, potassium biochromate, aqua ammonia, iodine, ammonium chloride, potassium iodide, and other reagents. Thus, the shortage of reagents was partly relieved.

On the basis of our experience, we suggest that comrades engaged in similar work around the country study chemical analysis literature, try using less chemical testing reagents than recommended, and make efforts to reclaim reagents from waste solutions. As long as we dare-to-think, dare-to-do, and overcome superstitions in the communist way of doing things, much can be accomplished. This type of work not only will solve material shortage problems but will also mean considerable cash savings to the country. Let us all move ahead together in establishing a socialist society at an early date.

II. INTEGRATED UTILIZATION PROBLEM OF A VERTICAL ELECTRIC FURNACE AND FURNACE GASES IN A MAGNESIUM PLANT

/Following is a translation of a portion of an article by Meng K'an-an in Yu-se Chin-shu (Nonferrous Metals), No. 8, Peiping, 18 April 1959, pages 34-35./

Up to the present time the production of metallic magnesium is still most commonly done by the chlorination electrolysis method, whereby the chlorination operation is carried out in vertical electric furnaces and the composition of the charge has a high content of carbon. The furnace gases of the electric furnace, as a result of contamination by the chlorine and because they contain large amounts of carbon monoxide, are very poisonous and must be scrubbed before discharging by a 120-meter high chimney so as not to damage the areas surrounding the plant and affect the health and growth of living things. Thus, not only is it necessary to install a series of gas cleaning and discharge facilities but also, much economic loss will result from the inability to utilize the high content of carbon monoxide. For these reasons, in the magnesium smelting book by the Soviet authors Ssu-t'eh-lei-lieh-tzu, T'ai-ch'ih, and Ku-liao-ni-ch'ih-chi, it was suggested that by employing a sealed method for the vertical electric furnace, whereby the concentration of carbon monoxide is raised to 25% in volume, a gaseous fuel with a heat value of about 760 kilocalories per cubic meter can be formed. When sealing is more effective, the carbon monoxide content of the furnace gas can be raised to 38% in volume and a gaseous fuel of 1,160 kolocalories per cubic meter will result; thus, for a magnesium plant with a daily capacity of 1,000 tons of metal, the heat obtained in 24 hours will amount to 35^{6}x 10 kilocalories. However, we know that in the operation of vertical electric furnaces, pressure conditions are normal so that sealing is a very good difficult proposition and most magnesium plants have not been able to effect this sealing. Another method of handling the problem is to make hydrochloric acid; for a 1,000-ton per day magnesium plant, about 145 tons of hydrochloric acid can be obtained in 24 hours; however, because of severe corrosion in the absorption equipment, this method is likewise uneconomic and difficult to carry out. Only in areas where HCL is very short will there be any justification of the method; thus, for usual magnesium plants, the method is not applicable. Other methods such as making magnesium chloride solution, etc., also have no economic and practical value. Under the leap forward conditions in our country's agricultural and industrial production, we must seriously consider the integrated utilization problems in industrial plant operations. Thus, in considering the design of magnesium plants, the problem of effective utilization of vertical furnace waste gases in conjunction with minimizing construction expenses is a very significant aspect.

For ordinary magnesium plants, particularly relatively large scale plants, auxiliary salt electrolysis units are often installed to utilize the cheap self-produced chlorine gas and, at the same time, produce large quantities of caustic soda; this method not only means better utilization of the chlorine and hence economy but also better safety with regard to chlorine storage. However, at present many salt electrolysis plants and units cannot find a market for the by-product hydrogen, and when not recovered, means losses. Such a situation should not be permitted. We must likewise consider the effective utilization of this hydrogen.

Under the leap forward conditions, the workmen and technical personnel strived to their utmost and finally developed the agricultural fertilizer ammonium bicarbonate, which requires low investments, means low cost, and is very effective; this new fertilizer has been most helpful to agricultural production. Thus, greater output of ammonium bicarbonate would directly affect agricultural development. If we can utilize the hydrogen gas from salt electrolysis units built in conjunction with the vertical electric furnace type of magnesium plants to make nitrogenous fertilizer, better resource utilization can result; this is an interesting idea which has definite possibilities.

III. NATIVE METHOD COPPER ORE BENEFICIATION

[Following is a translation of portions of an unsigned article in Yu-se Chin-shu (Nonferrous Metals), No. 9, Peiping, 10 May 1959, pages 18-26.]

The Yang-mei-shih Tungsten Mine native-method copper refining experiment work group was formed, under the leadership of the Party representatives at the Yang-mei-shih Tungsten Mine, by primarily the staff and workers of the Yang-mei-shih Tungsten Mine, comrades of the Kan-nan Metallurgical Bureau, the Kiangsi Metallurgical Research Laboratory, and the (nonferrous) section of the Kiangsi Metallurgical Bureau.

The Yang-mei shih Tungsten Mine has had considerable experience in hand concentration of ores, and since starting to refine copper in 1958, has also learned something about copper. With such a background and by learning from practices elsewhere, the staff and workers at the mine, in just a little over a month, built a native style copper beneficiation plant at Niu-ling and a native copper smelting plant at Hsia-lung, both of which have already started production.

We will describe our native method of copper ore beneficiation and raise some technical problems for discussion.

Importance of Copper Beneficiation

According to present knowledge, most of the copper ore in our province is low grade, generally under 1% Cu and sometimes only 0.5%. No direct smelting copper ore has as yet been found. In order to produce copper, ore concentration work cannot be avoided. This is also true of native smelting of copper.

The key to breaking the technical barier of native copper production lies in ore dressing or concentration. Once suitable copper concentrates can be made, the technical problems of smelting can be fairly easily solved. Concentrates containing more than 3-5% copper will be amenable to smelting; the higher the copper content, the easier the smelting. In some places, copper concentration has not been emphasized and low grade materials have been directly smelted; the result is much manpower and fuel expended without being able to produce much if any copper. We have had this type of sad experience in this province, as have others elsewhere. When material of less than 3% copper content is smelted, much copper is mixed in the slag which means high consumption of fuel, fluxes, and manpower, low recovery, and excessive costs. When 1% copper ore is concentrated to 5%, the total weight is reduced from 100 to 12; thus, much fuel, fluxes, and manpower can be saved, which mean lower cost, greater simplicity in smelting, and higher copper recovery. Although much manpower is needed in ore dressing, this will be more than compensated by subsequent savings in production, transport, fuel, fluxes, etc. By upgrading 100 metric tons of ore to 12 tons of concentrates before smelting, subsequent savings in fuel alone would amount to 40-50 tons. On the basis of this one fact, it becomes apparent why it is necessary to concentrate the ore.

Usually, by-product materials are associated with copper. Ore concentration not only helps eliminate impurities but also means the recovery of some by-products. Besides, a cleaner material facilitates smelting.

At our Niu-ling plant, we have made on an experimental basis 18 types of simple equipment. Wood is the main material used to make the equipment; no mechanical or electrical power is used although water and animal power are sometimes employed. Some of the equipment has become very effective, whereas some still must be modified and improved.

The Niu-ling native method copper concentration or beneficiation plant has been built on the basis of adding certain types of simple equipment to the tungsten mine's original manually operated gravity concentration plant. Niu-ling uses no mechanical power. Its flowsheet is as follows:

The Niu-ling tungsten-copper beneficiation plant treats 60 metric tons of mine-run ore per day. About 20-30 tons of waste rock are first thrown out by hand sorting. Ultimately 10-15 tons of copper materials are sent to the newly constructed copper concentration plant everyday for further upgrading. About 2-3 tons require treating by the palong-lauder method (similar to the practice of Chinese tin mines in Southeast Asia), and 5-6 tons require treating by flotation. Through these methods, more than 70% of the copper can be recovered.

Important Problems to be Considered in the Native Practice of Copper Ore Concentration

1. Emphasize hand sorting -- this is simple and effective. The more waste rock that is thrown away in early stages, the less the work of crushing and the greater the productive capacity.

2. Emphasize gravity concentration, "grain floating" (lauder principle), and flotation combination type of flowsheet. In copper concentration by native methods, the heavy type of crushing and grinding work should be employed as little as possible. Instead, the gravity concentration, "grain floating," and flotation combination type of flowsheet should be employed.

The ores at the Teh-hsing copper-lead-zinc mine are fairly well crystalized. A gravity concentration and "grain floating" combination method employed has shown good results with over-all recovery of about 63% of the metals (copper-zinc separation is still not too good.)

3. Selective separation of complex metal sulfide ores -- The flotation method is best. Gravity concentration can only separate copper-lead ores to a certain extent. "Grain floating" and flotation methods must be employed in separating copper-zinc ores, copper-molybdenum

ores, etc. Technical conditions are a little more complicated. Since copper is in great demand at the present time, copper recovery should be emphasized in beneficiating copper-lead-zinc ores.

4. In progressing from small native group to small modern group operations, the method of gradually adding mechanical equipment should be adopted — The experience of the tungsten mine is worth recounting: the first step was to add jaw crushing equipment and the second step was to install grinding equipment; much labor was saved and output capacity was raised. Concentrating equipment and transport (or conveying) within the plant can also be gradually changed from native to modern. During the Sou-wang-fen conference, the small scale mechanical concentration plant of 50 metric tons of ore per day and 70 horsepower motive power was introduced, which had relatively simple equipment. Conditions permitting, the native concentrating plant should be gradually coverted to the small mechanical concentrating plant.

5. Problem of ore beneficiation reagents -- Reagents should be self-made or made on a cooperative basis and substitute reagents should be infestigated. For example, the substitution of heavy oil for "Huang-yao" (yellow machine) /flotation reagent/ should be emphasized (actually, the use of heavy oil preceded "yellow medicine" in history of flotation). Experiments on the use of 112 grams of heavy diesel oil per ton of ore together with 90 grams of camphor oil and 1,000 grams of lime in floating a 0.523% copper ore to a 6.71% copper concentrate (tailings contained 0.093% Cu) at a recovery of 83.4% was successful on a preliminary basis; even better results might be expected by improving operating conditions.

At the Hsia-lung mechanical ore dressing plant, experiments have been successful in the use of pine shavings, leaves and cones to replace pine oil as the frothing agent in the grinding machine (i.e., ball mill); only about 0.2 kilogram or even less is needed to form good (not sticky and not brittle) bubbles. Soap can also be considered for use as a flothing reagent in copper, but conditions must be suitable and not too much soap should be used.

6. In native ore beneficiation, the grade of copper concentrates need not be too high-- Present tailings and other ground up materials should be saved for possible later use. The way to determine the suitable grade of concentrates in various copper producing areas should be based upon ore characteristics and beneficiating conditions. The "grain floated" product is usually high grade, often 15-20% or better. The gravity concentration and flotation products should be lower in grade (5-8%) so as to have better copper recovery. Any concentrate over 5% can be smelted. When ore separation conditions are extremely difficult, a minimum grade 3% product can be sent to the smelter.

The over-all metal recovery in native type of concentrators are of course not very high. Thus, waste rock and tailings should be saved whenever possible for later treatment.

IV. ORE DRESSING IN THE NONFERROUS METAL INDUSTRY DURING THE LEAP FORWARD

/Following is a translation of an unsigned article in Yu-se Chin-shu (Nonferrous Metals), No. 10, Peiping, 31 May 1959, pages 18-19./

Under the socialist construction main line, ore dressing work in the nonferrous metal industries has made great progress during the leap forward of 1958. A very good example is the problem of metal recovery, as illustrated by the statistics below:

Mine	One Dressing Method	Ore Treated	Mill Recovery 1956	1957	1958
Shih-tsui-tzu	Flotation	copper	98.84	96.16	96.75
T'ung-kuan-shan	Flotation	copper	90.89	92.92	93.79
P'an-ku-shan	Gravity Concentration	tungsten	82.43	84.99	88.70
Hu-k'ang	Gravity concentration	tungsten	75.73	81.70	88.4

During 1958 the mill recovery rates for all units were generally raised, particularly in copper and tungsten mills. For copper circuits, 6 units have mill recoveries exceeding 90%; and for tungsten circuits, 11 units have mill recoveries exceeding 80%. Among the ore dressing plants, the highest monthly recoveries during 1958 were as follows: T'ung-kuan-shan, 97.77%; Shih-tsui-tzu, 98.17%; and Hu-k'eng, 97.08%.

Next, the handling capacities for most ore dressing plants have been greatly raised. For example, for a certain lead-zinc mine plant in Central-South China, the daily capacity at the beginning of the year was 560 metric tons, and by the 4th quarter it became 1,200 tons for an increase of 1.14 times. The key to raising capacity lies in improving the efficiency of the grinding machines. Take the example of the Sou-wang for Copper Mine; the designed capacity for each unit of ball mill was originally 21 tons per hour, but during the third quarter of 1958 the tonnage handled was 28 tons and during the fourth quarter, 32 tons. Using the utilization coefficient (ore handled per hour per cubic meter volume) of the ball mill as the standard, the mines with this coefficient exceeding an average of 3 in 1958 included Shui-k'ou-shan, No. 1 Rare metals, and T'ung-kuan-shan.

In raising the handling capacity, all units adopted various measures, as for example: reduced the sizes of crushed ore from generally 10-12 mm to 6-8 mm; raised the rotation speed of grinding machines by 1-2 turns per minute; when appropriate, reduced the thickness of the lining plates as to raise the effective volume of grinding; improved screening and size classification work; emphasized "rational" loading of iron balls, etc.

The operational efficiency of grinding machines has been raised markedly. Among the flotation plants, 10 units have their ball mills working at better than 90% "Yun-chuan-lu" (conveying-turning rate). Among the gravity concentration plants, 10 units have their rod mills working at better than 80% "yun-chuan-lu." The turnover rates for the grinding machines of various mills were as follows during 1958:

Mine or Mill	Hua-t'ung	Fu-jung	T'ien-pao-shan	Huan-jen	Shih-tsu-tzu
Ball mill	98.60	97.40	96.48	95.75	95.62

Mine or mill	Hu-k'eng	T'ieh-shan-lung	Hsiang-hsi	Hsiao-lung	Ta-chi shan
Rod mill%	89.78	88.79	87.67	87.24	85.91

All producing enterprises have made many adjustments in their mill flowsheets. In flotation, the hydrotator has been commonly used in classifying and controlling the grain size, regrinding of middlings for further flotation has been stressed, the time for rougher and refined concentration has been lengthened, and for some units intermediate flotation is done. In gravity concentration, some units have employed grinding and concentration in steps and improved or strengthened flowsheets with regard to fine material.

Much has also been done in adopting new techniques and new equipment. The hydrotator has been widely used in many plants, resulting in better recoveries and higher handling capacities. Porcelain "whirling equipment" and rubber lined and "cast stone" hydrotators have been made successfully on an experimental scale by the Ore Dressing Research Institute and other organizations; their performance is better than existing equipment. The Humphrey spiral is widely used in our country's tin dressing plants; it is much better in recoveries and ease of operations compared with using wooden launders (palongs) alone. The spiral classifier made successfully on an experimental scale with cement, clay, cast iron, galvanized sheets, etc., by the Kwangsi Research Laboratory should be improved and pushed into wide use in the future. Many types of efficient gravity concentration equipment have also been in various places, as for instance the 20-section automatic launder, the double layer table, the 8-grade hydraulic classifier, etc., have all been widely used in production and are performing at 1-2 times greater efficiency compared with old equipment. Our country's mineral dressing research work has made great progress during 1958, particularly in oxide are beneficiation techniques. At the Kunming Metallurgical Research Laboratory and the Nonferrous Metal Research Institute, experimental work done under the guidance of Soviet specialists on the combination flowsheet for oxide ores was very successful. The process involves initial treatment by flotation to produce part of the copper concentrates, followed by hydrometallurgical treatment of flotation tailings. The recovery of the combination flowsheet reaches 85-90%, which is nearly twice as much as the recovery by flotation alone.

An item of experimental accomplishment which should be emphasized is the work of heavy medai separation or concentration on lead-zinc ores done jointly bh the Tung-pei Engineering College and Ch'ing-ch'eng-tzu mine. According to the experiment, ores 20-25 mm in size when treated by heavy media separation can be rid of about 50% of the waste rock; through adoption of this technique, the productive capacity for the ore dressing plant can be doubled and cost reduced by 35%, which means much savings and wealth for the country.

The Ore Dressing Research Institute has achieved experimental success on the treatment of scheelite middlings from the Yao-kang-Hsien Tungsten Mine by hydrometallurgical methods (high-pressure digestion and leaching). The digestion and precipitation rates are 98-99%, content of residues is 0.1-0.3% Wo_3, calcium tungstate (synthetic scheelite) produced is 63-65% WO_3, and the total tungsten recovery is 96-98%. On the basis of these experiments, this Institute, working jointly with the Kiangsi Metallurgical Laboratory, suggested a combination flowsheet consisting of gravity concentration and hydrometallurgical treatment which raises the tungsten recovery from the past 55% to 83-84%.

Much experience was also gained in native milling methods during 1958. Many simple types of milling equipment and beneficiation reagents were made in various places, particularly in the field of copper with regard to the design of native-modern combination small mills. The way to rational development of small native group mines has been found.

In reviewing the accomplishments for the last year, we realize that this year's responsibilities are indeed glorious and great. Compared with last year, copper and lead output are to be more than doubled and large increases are also planned for other nonferrous metals.

To further develop the latent potential mf minieral dressing performances and ensure the completion of targets, we require on the basis of the great leap forward of last year to push the mass movement centered around the revolution of techniques. Under the leadership of the Party and by fully utilizing the strength and intelligence of the masses, we must attain the following standards and attain an even grester leap forward of 1959.

1. Raise the grade of concentrates, particularly in copper and as much as twice previous grades.

2. The utilization rate of grinding machines should be raised 30-50% above past rates.

3. Metal recovery rates should be 2-8% better than during 1958.

4. The "Yun-chuan-lu" (turnover rate) for milling equipment should reach 95-98%.

V. COMPARISON BETWEEN 5,000-AMPERE CONINUOUS ANODE TYPE OF ALUMINUM ELECTROLYSIS CELLS WITH THE ORDINARY NON-CONTINUOUS TYPE

/Following is a translation of portions of an article by Han T'ien-yu, Sung Wei, and Li Pao-hai in Yu-se Chin-shu (Nonferrous Metals), No. 10, Peiping, 31 May 1959, pages 19-21./

The Peiping Aluminum Experimental Plant was partly placed in production during September of last year. As of now, 39 ordinary non-continuous type of anode aluminum cells and 13 continuous /Solderberg type/ cells are in operation. The object of this paper is to compare the two types of cells on the basis of the experience at the Peking plant for reference in construction of other small plants around the country.

A. Comparison from the Technical Viewpoint

The 39 non-continuous type of cells started to operate in September and by November they were working on a normal basis! The 13 continuous type of cells started to operate in December and January they were working on a normal basis. The two types of cells are compared on the basis of February and March data. It should be noted that the technical indices are rather low in February because part of the cells were not yet stabilized. The various technical indices are listed below in Table 1:

	Current Efficiency		Cell Voltage(1) (volts)		Electricity Consumption (kwh per mt of aluminum)	
	February	March(2)	February	March	February	March
Continuous cells	57.8	86.5	8.00	7.36	43,000	26,100
Non-continuous cells	55.2	67.5	7.93	8.15	44,400	37,200

Note: (1) Electrolysis consumption coefficient calculated on the basis of working voltage plus 0.3 volts.

During March, trouble developed in the rectifiers and power source: electricity was stopped 4-5 times (each time more than 15 minutes and the most serious time was three hours and 35 minutes).

(1). Electrolysis consumption coefficient calculated on the basis of working voltage plus 0.3 volts.

(2). During March, trouble developed in the rectifiers and power source; electricity was stopped 4-5 times (each time more than 15 minutes and the most serious time was three hours and 35 minutes). Each power failure had some effect on electrolysis, hence, if there were no stoppages, the current efficiency would be somewhat higher.

From the above table it is clearly seen that the continuous type of cells perform much better than the non-continuous type of cells. This is particularly obvious in the technical indices for March.

B. Comparison From the Economic Viewpoint

As for an analysis from an economic viewpoint (whether it be the basic construction cost of the cells or the subsequent raw materials consumption and production cost), attention on details was not paid on the matter so we can only give some general observations.

1. In discussing the disadvantages of the continuous anode method, it should be said that the construction of the anode is more complicated and therefore requires a little more time and steel and copper materials. According to calculations for each self-baking anode cell compared with a non-continuous cell, the requirements have been increased 200 kg for steel materials and 10 kg for copper materials. Therefore, for a 1,000-odd ton per year plant with 100 self-baking cells, the additional materials needed over the similar number of non-continuous cells would be 20 tons of steel and 1 ton of copper. The added investment necessary for the self-baking anode cell will be more than compensated in lower operating cost.

2. Comparison of raw materials consumed between the two methods on a unit bell basis.

Although this statistical table cannot be considered standard, it can be readily seen that that whether it be electricity consumption or the various items of raw material consumption, the self-baking anode electrolysis call is superior than the non-continuous cell. Conservatively, we guess the followwing economic results:

(1) For every metric ton of aluminum, savings in electricity would be 5,000 kilowatt-hours. At the price of 7.22 "fen" per kilowatt-hour, total savings would be 360 yuan.

(2) Savings in cryolite would be 50 kilograms, or, at 1,180 yuan per metric ton, 59 yuan. Savings in aluminum flouride would be 5 kilograms or, at 1,650 yuan per ton, 8 yuan. The two items aggregate 67 yuan.

(3) The cost of anode paste, calculated at 570 kilograms (383 yuan per metric ton price) would be 220 yuan; and the cost of carbonaceour materials, calculated at 800 kilograms (921 yuan price per ton) would be 738 yuan.

(4) The self-baking anode cell requires aluminum plate-- six kilograms cost 33 yuan.

(5) It is difficult to calculate the depreciation charges for the self-baking anode cell over the non-continuous cell, but we think it would not exceed 50 yuan.

Summary savings would be 360 plus 67 yuan plus 510 yuan minus 33 yuan minus 50 yuan, or on the order of 854 yuan.

In other words, the aluminum produced by the self-baking anode method would be about 850 yuan cheaper per metric ton than the non-continuous method. We feel this estimate is conservative. We have not included the item of wages, although we know that less labor is involved in the self-baking cell method. For a 1,000-ton per year aluminum plant using the self-baking method, the savings per year over the non-continuous method would amount to 850,000-odd yuan. If we accept the March statistics as being representative, then yearly savings would exceed 1,200,000 yuan.

The above discussions fully show the advantages of the self-baking anode method over the non-continuous method. If we can solve the steel and copper material supply problem, then we certainly can push the adoption of the self-baking anode electrolysis cell method.

VI. EXPERIENCES ON MODIFYING FLOWSHEET, DEVELOPING LATENT CAPACITY OF EXISTING EQUIPMENT, AND CONVERTING ONE PLANT TO THREE PLANTS

/Follwing is a translation of portions of an unsigned article in Yu-se Chin-shu (Nonferrous Metals), No. 11, Peiping 11 June 1959, pages 16-18./

During the leap forward of 1958, after much hard work and changes in production quotas, the final output of the Ta-ch'ang Tin Mine) roughly doubled the 1957 rate. At that time, expanding ore extraction was not a severe problem; the key to greater production depended upon the ability to raise the ore dressing capacity.

The best method to raise the ore dressing capacity was to develop the latent capacity of existing equipment, modify flowsheets, and increase the ore handling capacity. Under the correct leadership of the Party and through hard work by the staff and workers, with small investments and a short period of reconstruction, the milling capacity was raised to 300% of the original, which meant in effect that one plant was converted to three plants.

In the expansion program carried out, investments amounted to only 67,000 yuan and the plant was stopped for only a week. This compared with the program suggested by the Ch'ang-sha Design Institute which called for an investment of 700,000 yuan and work stoppage of more than a month. One basic deviation from the Ch'ang Institute plan is that the plant building was not reconstructed.

Conclusion

The mill was ultimately expanded to three times the original capacity. Production targets were achieved in conformity with the Party's policy of "more, fast, good, and economical."

According to present operating conditions, we need to further investigate and solve the following problems:

1. Although jig concentration has been very effective, jig tailings are diluted because the pulp entering the classifier contains much water and within the classifier this water cannot be easily raised so that the classifying action is hindered. This also affects tabling operations. We are contemplating changes but, limited by plant floor space and inadequate grade or slope, find it difficult to make the changes. Of course, the best flowsheet would be to classify first and then jig; after the jig tailings are concentrated, classify again and send to the tables for separation.

2. The crushing problem has not been adequately solved. At present the chrushing time is too long (more than 20 hours each day) and storage capacity is relatively small. If there are any unforeseeen incidents or obstacles, grinding may be affected and stoppage may result. Moreover, grain size control is not too good and pieces are sometimes too large.

3. Recovery from slime is not too good since these contain too much tin.

VII. INVESTIGATIONS AND DISCUSSIONS ON THE MILLING OF FINE TUNGSTEN ORE

/Following is a translation of portions of an article by Chang Kuo-p'ing in Yu-se Chin-shu (Nonferrous Metals), No. 11, Peiping 10 June 1959, page 19-21./

Since tungsten ores are very friable, much fine ore or slime in sizes smaller than 74 microns are formed in mining, crushing, and milling; this is difficult to recover and results in large losses in metal content. Since 1956, the various tungsten mines in Central-South China have adopted many measures to handle fine tungsten, using hydrotators, automatic launders, or classifying equipment to solve the roughing concentration problem of fines. However, the quality of the up-graded tungsten fines is only 17-20% WO_3 and with much impurities so that the product cannot be marketed; further concentration is difficult because of the fineness of the material. In the past, the product has been either made into synthetic scheelite or mixed in small proportions with high quality concentrates. The former is a hydrometallurgical method requiring large amounts of soda ash or caustic soda; operations are complicated and costs are high. The latter is limited by the amounts that can be mixed. Both methods are not ideal.

To solve the further concentration problem of fine tungsten, much work was done in the fourth quarter of last year, under the leadership of the Party and close cooperation between the technicians and experienced workers, to find a suitable flowsheet for these fines. By using a combination method involving flotation, grafity concentration, and roasting, within two months the fine tungsten intermediate product accumulated in a year was concentrated into several tens of tons of high grade concentrates that could be mixed in large proportions into final products meeting specifications.

Conclusions

Milling of tin-tungsten ores is characterized by complicated equipemnt and flowsheets not only with regard to large grain size products but also fines. Treatment of fines is even more difficult because the mass is less, total surface area is greater, and the force of attraction between grains is greater so that concentration is more difficult and the adoption of combination flowhheets is natural. This explains whay in the investigations and production in our plant we chose the flotation-gravity concentration-roasting combination method to upgrade tungsten fines. However, this type of flowsheet lacks magnetic separation cycles so that the finished concentrate produced contains high tin.

To further improve the problem of concentrating fine tungsten materials, it is necessary to develop a combination flowsheet involving four types of operations -- flotation, gravity concentration, magnetic separation, and roasting. Among the techniques related to this combination flowsheet, further investigations must be made in the magnetic separation aspect. For example, experiments are being made to use "surface" active materials" to prevent sticking between the grains with a view to making entirely satisfactory tungsten concentrates.

VIII. USE OF REVERSE BLOWING FURNACE FOR SMELTING COPPER-IRON ALLOYS

/Following is a translation of an unsigned article in Yu-se Chin-shu (Nonferrous Metals), No. 12, Peiping 18 June 1959, page 24./

The reverse burning furnace is a smelting furnace connected with a combustion furnace. There is a fire duct in between which depends on a draft for conveying the flame. The furnace is shaped like a round mount.

A. Structure of the Furnace

1. Smelting furnace: The furnace base with cross-like air ducts is first built to make the furnace moisture-proof; this base is constructed with stone or blue brick. The furnace bottom and furnace walls are built with refractory brick, the bottom being 20 cm thick and carefully sealed to prevent the leakage of molten copper. The furnace top is constructed with old kerosene drums cut at a height of 20 cm, inside of which are refractory materials. At the furnace bottom, 20 cm high there is a copper discharge opening; at the center, 40-cm high, there is an operating gate for charging materials; to avoid "poisonous gases" consisting of coal and sulfur gases, a 3-meter high and 25-30 cm diamete chimney (or flue duct) should be constructed on the top part of the operating gate; finally, at the upper central part of the furnace, an oxygen tube or pipe is installed for adding oxygen and ridding sulfur gases.

2. Combustion furnace: It is best to use refractory bricks to build this, but blue bricks will do. The walls of the furnace "stomach" or hearth should be built 10-16 cm thick with fefractory materials; a slag discharge opening is left in the bottom part of the furnace. An air intake opening is left at the bottom of the grate, and a coal ash removal opening is left above the grate. The top of the furnace is similar to that of the smelting furnace, with however a 20-cm diameter opening left in it for charging coal. This opening has a removable lid, and should be sealed when coal charging is completed (any leakage may mean that the top will melt).

3. When construction of the smelting furnace reaches the height of 50 cm, a flue duct is left open. For the combusion furnace, the height for the flue duct should be 70 cm. The flue duct between the two furnaces should be at an angle of 50 degrees; it should be made of refractory brick, sealed carefully and, after the furnaces are completed, enclosed by iron frames to avoid cracking.

4. According to data on a reverse blowing furnace built at our plant and capable of handling 2,500-3,000 chin of copper matte per day, the construction cost is 700-900 yann (includes labor and materials). The dimensions of the furnaces arw: for the smelting furnace, the inner diameter is 110 cm and the height is 80 cm; for the combusion furnace, the inner diameter is 80 cm and the height is 140 cm.

B. Equipment Related to the Furnace

1. Using a 12 horsepower internal combusion engine, two units of reverse blowing furnaces can be simultaneously supplied with air from a blower connected to the engine.

2. Five-men per shift operate the funaaces--two "in front of the furnaces" and two operating equipment.

3. Tools in front of the furnace: a balance, three shovels, three iron rods (with point) two slag or residue rakes, one long ladle one long rod, etc.

C. Operational Procedures

1. After all preparatory work is completed, wood is used to light the fire, some lump coal is added, then fine coal (better lump all the way) after the lump starts to burn, and a draft is quickly introduced. After the lid of the combustion furnace is sealed, wait for the fire tin the smelting furnace to get red, then charge in the copper matter all at one time. This is followed by adding a 4 to 6 mixture of coke fines and clay. Half seal the operational gate in line with the bottom of the fire opening or a little higher so that the molten copper matte will not flow out.

2. After the copper matte is melted, add quartz so that iron sulfide slags form iron oxide. At this time, use an iron rod to test the depth of the slag. According to the depth of the slag, make small troughs outside the operating gate for slagging (if slag does not come out, use a rake to help). At the same time, use the long ladle to take out some molten copper for analysis to see whether it meets standards (more than 30% Cu); if not, add more quartz until the standard is reached. In front of the copper opening, build a copper trough; the copper matte can be "blown smelted"; t molten copper can be tapped once every 12 hours.

3. According to our plant"s experience in "blowing smelting" copper matte, 7-15% materials can be raised in grade to 30% plus with 20-25% quartz and 20-25 materials, with about 15% quartz.

4. On the basis of "blowing smelting" copper matter, one can similarly advance one step and make crude copper. The first step of smelting crude copper is the same as for copper matte. In the second st step, air is blown into the melt so that the iron sulfides are oxidized to form oxides and sulfur dioxide. The iron oxides come out as slag and the sulfur dioxide leaves with the smoke. When the copper matte contains very little iron, "white copper matte" is formed. According to the tested amount of iron, the addition of 30% quartz is adequate for making suitable slag. In the third step, a pure copper sulfide (white copper matte) is converted to copper oxide. Unconverted copper sulfide makes a little slag. Finally, add wood or wood shavings into the molten copper to reduce copper oxide to metallic copper.

D. Performance

1. Low quality copper matte can be made into satisfactory grade matte.

2. Iron-copper alloy can be smelted into crude copper.

3. Crude copper can be made from satisfactory grade copper matte. For example, on 16 April during 27 hours of copper extraction, 2,000 chin of 20% copper matte, 1,100 chin of fuel (mine-run coal), and 250 chin of quartz were consumed and 70 chin of crude copper of 90% grade were drawn out.

E. Items of Special Attention

1. Preparatory work must be well done. The furnace must be very dry before firing. The smelting furnace bottom will stick and the combusion furnace will crack if not well pre-dried.

2. In testing slag, special care must be taken not to introduce water or something cold into the furnace test explosions result. In tapping copper, the troughs must not be too wet to prevent the matte from expanding and burning and causing losses in copper.

3. During operations, workmen must wear glasses, mouth coverings, and other safety equipment because of gases and possible accidents.

4. After copper is tapped and the opening is about to be sealed, the opening must be first cleaned, and patched up inside and outside so that no molten copper will flow out by itself.

5. Coal must be added according to the color of the fire at the opening. When the color turns from white to red-black, add coal. The draft duct should be first closed, and coal loading should be done quickly. The lid is sealed once more, the draft opened, and normal production resumed.

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